CEMENT AND LIME MANUFACTURE

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Mills for Clinker Grinding.

(CONTRIBUTED).

THE practice and procedure of clinker grinding have possibly passed through more changes than any other preparatory process, but finality has not yet been attained. It is unlikely ever to be attained, for there will always be a call for better quality, for the process to be carried out in less time, for the process to be more self-contained, and so on. Moreover, every step in one direction prompts thought to other phases. For example, improvements in grinding may indicate that the grinding could be more efficiently carried out if crushing were more effectively performed, and so more attention would be given to crushing as a means of obtaining higher efficiency in grinding. As far as crushing is concerned, an effort might be made to obtain a more uniform size of product, or an increased quantity of smalls, or a general smaller size of the whole, etc. In the meantime the need for a more consistent size of mill product might be called for, or a more uniform rate of decrease of particle size.

The fundamental procedure of wet and dry grinding is the same although the details of the mills differ. Materials of the same characteristics and of consistent hardness will call for substantially the same amount of power whether ground wet or dry, provided all other conditions are the same. One advantage with the wet process is that the mix will pass through the mill with the minimum head and the smaller particles will flow forward with the liquid automatically, whereas under dry conditions the grit, or " fines," will require a greater head to ensure passage, and the head necessary must be produced within the mill.

Over a long period it was the practice to use primary and secondary crushers for the preliminary part of the clinker grinding process, and mill-stones with sieves for the finishing part ; the sieves limited the size of the largest particles in the finished product and the size and the amount of the smallest particles were determined largely by returning the sieve oversize to the stones for further reduction. No complete classification of the product is now available, but the

practice appears to have resulted in a satisfactory decrease in particle size and a sufficiency of " fines " ; it is unnecessary to add that the sieve opening was much larger then than it would be now.

The capacity of the separate mill-stone units was small. The stones also required frequent dressing and running adjustment, which entailed frequent stoppages. A design of ball mill with sieves was later evolved, but as the particles of grit were discharged as soon as they reached the size of the hole in the sieve, the mill product contained only a small percentage of fines—it was actually gritty. This in turn prompted the development of the tube mill as a finishing element. This was all to the good, as it enabled the sieves to be used at an intermediate part of the process, but sieves and sieve changing were regarded as a nuisance owing to the frequency of changing the sieves and the loss of time that resulted from it.

In trying to avoid the use of sieves centrifugal separators and classifiers were experimented with, but they did not prove altogether successful. Separators, without any means of adjustment, were frequently used as traps for over-size pieces of grit ; the size or weight of the largest or heaviest particles in the fines would be determined by the characteristics of the separator or by the speed at which the revolving part rotated. The amount of feed to the separator would be 10 per cent. or 20 per cent. greater than that of the new feed to the mill, and this quantity reappeared as over-size at the separator discharge and was taken back to the mill feed.

Classifiers were usually fitted with some adjustable element which would enable an amount of control to be obtained over the size or weight of the largest or heaviest particles in the fines. Classifiers were frequently arranged for dealing with two, three, or several times the weight of the new feed to the mill. The greater the amount of the circulating load the smaller the amount of excess grinding on the finest of the whole product and the fewer the number of small particles produced, owing to their shorter period of grinding—the product could, of course, be gritty.

Several conditions operated against separators and classifiers, such as (1) The varying weight and density of the material, (2) The abrasive nature of the material resulting in wear and out-of-balance of the rotating parts, and (3) The absence or low percentage of really small particles in the finished product.

Occasional units of this kind are still being installed to meet special conditions, but there is no indication that they will be universally adopted, in fact the indications are against that possibility. They serve a special purpose when a product containing the minimum amount of fines is required and when it is possible for the mill to carry a circulating load several times the weight of the new feed, and under such conditions the reduction in size of the grit particles in a single pass through the mill will be very small. In this way the product of the mill with separator will contain no large particles owing to the selective characteristic of the classifier. Neither will it contain a large quantity of fines owing to the short period the grit has remained in the mill and the small amount of

grinding to which it has been subjected ; the small amount of fines it contains may impose some limitation on the use of the product.

During recent years a number of separators or classifiers has been installed as a means of increasing the capacity of the milling section of existing plants ; this may have proved satisfactory where the quality of the product is determined by the sieve residue and not by surface area. Surface area, however, can be increased to some extent if the mills and their equipment are arranged for carrying a large circulating load.

Coming now to the use of tube mills as a finishing element, the earliest mills were about 4 ft. in diameter by 16 ft. long. They were charged with egg-shaped flint pebbles, hand picked from the sea shore ; the usual diameter was 1½ in. to 2 in., and the length was rather greater than the diameter ; in bulk they weighed about 100 lb. per cubic foot. The mills were first lined with small glazed tiles and later with silex blocks about 2 in. or 2½ in. thick, cemented in. The use of flint pebbles, but of smaller dimensions, continued but the size of the mills was increased to about 6 ft. diameter by 26 ft. to 28 ft. long, but the demand for an increasing fineness of product indicated that the pebbles were too large and that media of greater density was necessary. To meet this requirement a short length of the mills was partitioned off at the outlet end and charged with metallic media of smaller dimensions and greater density. This media was of cast iron or mild steel, usually ¾ in. to 1 in. diameter and 1 in. to 1¼ in. long ; in bulk it weighed about 260 lb. per cubic foot. When the media had become conditioned the product of the mills became finer and the output substantially increased.

This increased and finer product was not obtained without cost, as the power absorbed by the mill increased pro rata with the increased weight of the media. However, as the mills were not designed for this considerable increase in weight it became necessary to linit the total weight of media used. Ultimately, many of the original mills were charged exclusively with metallic media having a total weight about 1½ times that of the pebbles superseded ; this in turn called for an increase of 40 per cent. or 50 per cent. in the power required for driving the mill, and the output increased in approximately the same ratio as the power absorbed.

The demand for larger mill units resulted in the economic limit in the size of the mill being reached, owing principally to its heavy and costly construction in the larger sizes. Hitherto the ratio of the power taken by the ball mill and the tube mill had been, say, 30 per cent. to 40 per cent. and 60 per cent. to 70 per cent. respectively, and 120 h.p. appeared to be the economic limit for the construction of the sieve-fitted ball mill. But no such limit controlled the design of the tube mill, and as experience had already been obtained with two-chamber mills a modification of the existing design produced the larger tube for the compound mill which could be equipped as a combined ball and tube mill and thus become a self-contained unit.

While these developments were taking place other preliminary mills and single-mill units were also developed, including the single-roll Griffin, the Giant

Griffin mill, the Fuller mill, the Hercules mill, and other mills of the same general type; these, or most of them, were of the high-speed type. Work was also put into slow-speed mills, including the rod mill, the balanced mill, and the Hardinge mill, as well as modifications of these and other types.

The general, and serious, objection to all the high-speed mills was that of vibration and wear, both of which became intensified as soon as wear took place; it is even possible that these characteristics prevented these mills from being constructed in the larger sizes. Some of the difficulties appear to have been got over in the case of the Hercules mill; in any case this mill appears now to be constructed in the 30-ton or 35-ton per hour size as a preliminary unit or for preliminary duty.

The rod mill (balanced) is essentially a preliminary mill or a grit-producing unit, but it does not appear to have been constructed in the larger sizes. The mill is of heavy and costly construction; grinding is effected by the action of several masses of heavy steel bars in rolling engagement with each other and not by impact as in the case of the ball and tube mills; the bars are made of hard and very tough alloy steel; and the grit passes between them in the process of grinding. The balanced (tube) mill is also of heavy and costly construction and is essentially a finishing mill. It appears to be of limited capacity for its size, although its power requirements are very low. Grinding is effected by the rolling or trundling action of masses of small balls (or equal) in radial chambers, and not by impact as in the case of the normal tube mill.

The Hardinge mill is a single-mill unit, a section through which is shown in Fig. I; the diagram also shows the profile of the charge under working conditions. The mill is pear-shaped the feed enters at the large end and the product is discharged at the small end. It is claimed for this mill that the coned-shaped portion at the discharge end has a segregating effect upon both the media and the material, and that the largest media and the largest grit congregate at the feed end and the smallest of both at the discharge end. As this mill is a single-mill unit the media may vary in size from, say, 3 in. down to the smallest size used, the latter depending upon the fineness of the product. This segregation probably only occurs if the charge volume and the amount of feed are both kept low and the latter is kept well under control so that the particles have to "climb" to get out of the mill. If this is so, it is important that this mill should not be over-fed. If the speed of the mill is correct for the larger diameter it is difficult to see how it can be so for the smaller diameter, when the smaller media may not be operating at its best speed, and the work done may be by attrition and not by impact. To obtain efficient grinding in any mill it is necessary for the mill to run at the speed suitable for the diameter, and this seems impossible with a coned mill. This mill is very popular under wet conditions, for which a large number are in use.

Reference may now be made to the compound or multi-chamber mill, which is essentially a single-mill unit of the slow-speed type. The details and arrangement of the mill have passed through many phases and it is now possibly the

most popular type of mill in service for dry duty. It is probable that at the present time the greater proportion of new mills under consideration or under construction are of the multi-chamber type. This mill can be constructed equally well in both the smaller, say 50 h.p., and the largest, say 1500 h.p., sizes respectively, and there would be no difficulty in making it larger if suitable conditions and duty could be found. The 1500 h.p. mill will produce, say, 40 tons and 25 tons per hour of ordinary cement and super-fine cement respectively; similar figures for the smaller mill would be, say, 1¼ tons and ¾ tons. The total output during a 140 hour week would be, say, 5,500 tons and 3,500 tons and, say, 175 tons and 105 tons each in the order given. Needless to say the larger mill should be the more efficient unit for cost-per-ton and should yield the most consistent product.

A distinct advantage of the single-mill unit is the absence of elevators or conveyors (or both) during grinding; the mill can be fed direct with the clinker

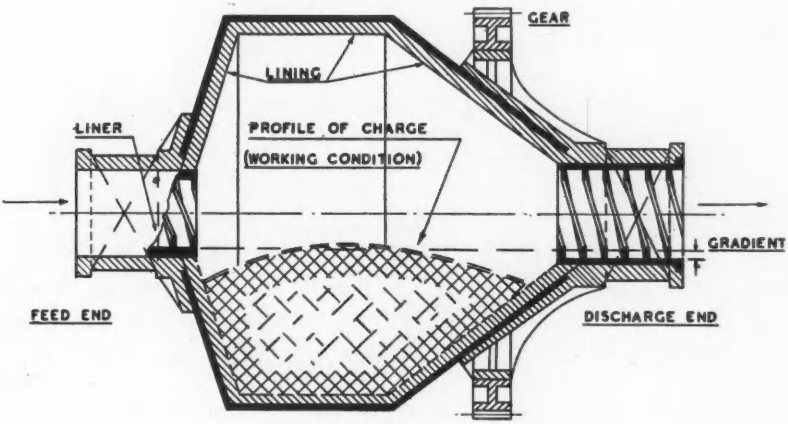

Fig. 1.

produced by any normal rotary kiln, provided the nodules are not greater than ⅜ in. to ½ in. diameter, and produce a commercial cement of any standard fineness, or surface area, in a single process.

Originally there were difficulties, owing to the heat generated during grinding, in obtaining a media classification suitable for grinding to normal and superfine cements without adjustment, with flaking under fine-grinding conditions, and owing to "nibs" being discharged with the product. Wider experience with this mill has enabled all these difficulties and others to be got over; the characteristics of the mill are also much more fully understood. Assuming the media classification to be correct it is possible to obtain from this mill a product wherein the particle size is of a consistently decreasing dimension from the maximum as determined by the sieve to the smallest size the media is capable of producing. The classification of the product is quite different from that ob-

B

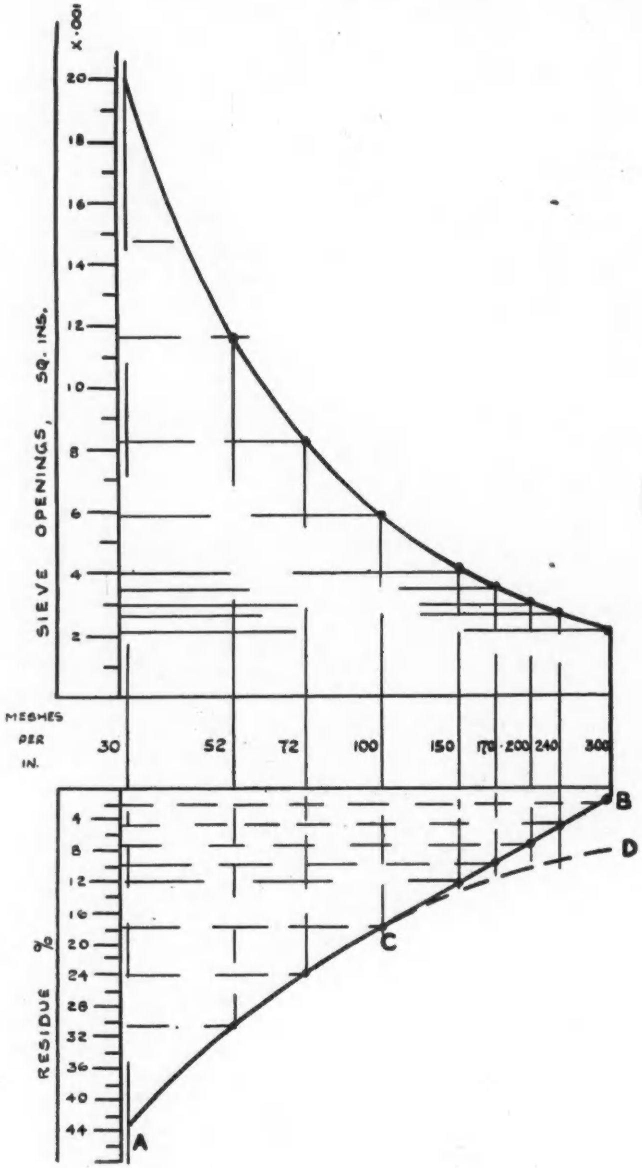
× ·001
SIEVE OPENINGS, SQ. INS.
20
18
16
14
12
10
8
6
4
2
MESHES PER IN.
30
52
72
100
150
170
200
240
300
RESIDUE %
4
8
12
16
20
24
28
32
36
40
44
A
B
C
D

Fig. 2.

tained from the mill-with-separator combination operating with a circulating feed of any normal ratio.

Fig. 2 indicates the characteristics of the cement produced by the compound mill charged with media of commercially correct classification; it also indicates how cement produced by a similar mill but with separator may be compared with it. The upper curve shows the connection that exists between the number of meshes per inch in the sieve and the dimensions of the sieve opening. The lower curve (AB) shows the connection that may exist between the residues obtained on a range of standard sieves from samples taken from a compound mill charged with media of commercially correct classification; Line CD indicates the influence a classifier would have upon the product from the same mill; the location of the point C and the height of BD will be determined by the classifying effect of the classifier, and also upon the amount of oversize being returned to the mill. Curve AB, which is the mean of a number, may deviate from that shown owing to a different classification of media or even to a change in the amount of feed being given to the mill, but any change can be explained or corrected for in the adjustment, feeding, or charging of the mill.

[With reference to our contributor's comment on the Hardinge mill, International Combustion, Ltd., state that the mill indicated in Fig. 1 is not used as a fine grinding unit by itself, but the conical shape causes segregation of the mixed charge, so that the grinding media with greatest drop act on the coarse feed, effecting a rapid reduction; the cone accelerates the discharge of those particles which are already fine enough. When a fine product is required the mill is operated with dry or wet classifiers as the case may be. The company continues: "The author refers to the many wet grinding installations; but the grinding principle is the same for dry reduction. It may therefore be of interest to record that during the past fifteen years dry grinding installations for various industries have exceeded those installed for wet grinding."]

Lime Solubility of Portland Cement.

AN investigation by R. Zollinger has been carried out in connection with the selection of a cement for concrete exposed to the action of water containing free carbon dioxide, and the results have been reported in "Zement." According to "Building Science Abstracts," the twelve test cements included Portland cement, Erz cement, iron-Portland and Portland-blastfurnace cements, trass cement containing Portland cement clinker and trass in the proportions of 70 : 30 or 60 : 40, cement with added "Thurament," and artificial trass. The problem studied was the extent to which the protective influence of the reactive silica added to the cement results from chemical reaction as distinct from the physical effect of increased density due to the colloidal volume increase. The latter effect was excluded by the use of porous test specimens into the interior of which the water could penetrate. The cement-sand (1 : 5.5) specimens were prepared in the form of slabs, 20 cm. by 20 cm. by 2 cm., and were exposed in separate glass tanks to a continuous flow of water containing carbon dioxide which was changed when the lime content exceeded 0.2 g. per litre. At intervals during a period of 210 days the quantity of lime dissolved from the slabs was determined from analysis of the tank water. On the conclusion of the experiments the specimens were removed from the water and, the disintegrated material having been brushed off, were stored in damp air for seven days. The specimens were then weighed and the loss of weight obtained. The loss of weight of the residual slabs exceeded the weight of the removed disintegrated material ; the amount of the additional loss of weight was an indication of the extent to which the slabs had been attacked. The data of loss of weight and lime dissolved and the lime solubility curves of the test cements are shown.

The tensile strength of the test slabs at the end of the test period of 234 days and of other slabs at the end of the same period was determined by bending strength tests. The strength test results showed not only a reduction of strength in each case but also a complete rearrangement of the strength values of the different cements. A second series of experiments was carried out with slabs of the same cements at the end of a storage period of six months in damp air to allow time for a chemical reaction similar in nature to the protective reaction to take place.

The results obtained with each test series are considered and the test cements placed in order according to the degree of corrosion, lime solubility, and effect on tensile strength. A decisive statement respecting the value of the individual cements, on the basis of the experimental data, it is pointed out, would be misleading. It may be said definitely, however, that in no case is the chemical protective action adequate to prevent an attack by corrosive waters. Chemical reactions with protective effect undoubtedly take place. The age of the concrete is not without importance, but complete protection is not attained with increasing age, and different types of cement differ considerably in their behaviour. Where, in practice, resistance to the action of corrosive water is observed it is due in part to the physical waterproofing effect of the colloidal compounds.

Mill Lining for Classifying Media.

A TUBE-TYPE mill, 8 ft. diameter by 24 ft. long fitted with a conical lining, in sections, for use on clinker grinding, is shown in Fig. 1. This is reproduced from " Rock Products " for April, 1938, which also dealt with smaller mills and models to which the same general type of lining had been applied. A reference was also made to this lining in Technical Paper No. 581, " Ball Mill Grinding," issued by the United States Bureau of Mines in 1937.

The efficiency of all compound mills is determined largely by the use of media of correct size, segregated in such a way that the largest balls impact upon the largest nodules of clinker and smallest media upon the smallest particles of grit. It is impossible to achieve this under most conditions, but anything done in this direction will prove of value. The normal compound mill is usually divided into three, four, or even five chambers by means of diaphragms—the actual number may depend upon the diameter and length of the mill shell. The media used are progressively smaller in the successive chambers from the feed end, and

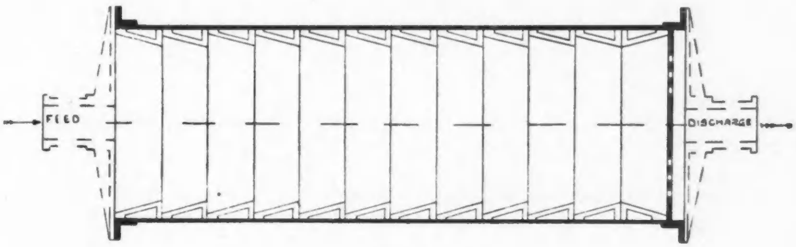

Fig. 1.

the clinker and grit become progressively smaller in passing from the feed end of the mill to the discharge end.

The energy of the large balls would be largely wasted if they were used on the grit towards the finishing end of the mill, and the energy of the small media would be non-effective if used on the nodules of clinker at the feed end of the mill. Keeping the large media and clinker nodules together, and the smallest of both together, operates to the advantage of the whole process. The separate chambers of the mill and the select loading of the media contribute to the desired segregation, but they do not go far enough—they are too " steppy "—and there appears good reason for assuming that the coned lining shown on the diagram will prove more effective as it would enable the size of both the media and the grit to be consistently progressive.

It is not practicable in the compound mill to provide a ball of optimum weight for each clinker nodule and particle of grit, but it is possible with the multi-chamber mill to arrange for a group of balls of a limited range of diameters to

operate upon a mass of clinker of general large size at the feed end of the mill and of smaller size at the finishing end ; this is the principle upon which the operation of the mill is based, but even so, the sizing of the media will not be of a consistently decreasing size ; the more natural merging of the sizes in the mill fitted with cone lining appears to promise better results. Apart from the steppy characteristic of the media in the compound mill the diaphragms cause inconvenience ; diaphragms are costly, some of the parts have to be frequently renewed, the separate chambers call for additional manhole openings with fittings, and strengthening plates have to be applied to the outside of the shell. Much of this inconvenience can be eliminated by the adoption of a coned lining.

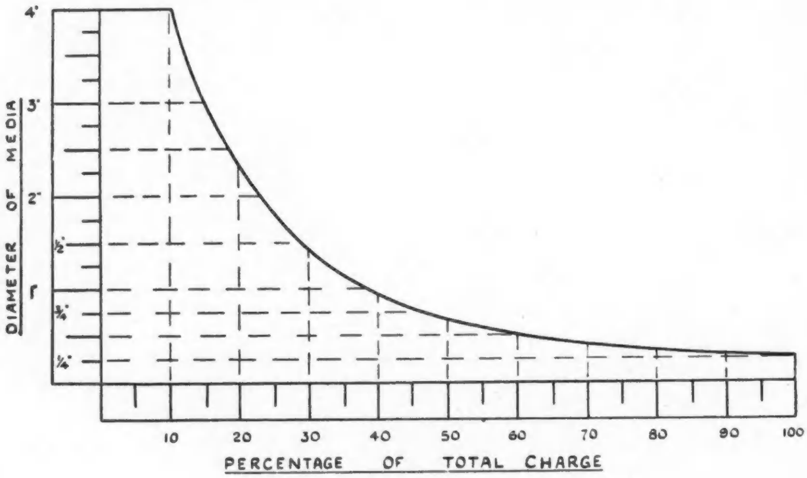

Fig. 2.

The rule for the grading of the media for the compound mill was given in detail in our issue for May, 1942, and is as follows :

$$D = 16d + \tfrac{1}{4} \text{ in.}$$

when D and d are the average diameters in inches of the media and grit in the respective chambers of the mill. The rule is applicable when the diameter of the clinker nodules is not greater than ½ in. and the particles of grit in the last chamber are, say, between 0·05 in. and 0·005 in. ; this range for the last chamber is largely assumed or estimated, as there are no exact figures available.

The weight of balls varies as the cube of their diameters, that is to say, the weight of one 4-in. ball is equal to that of eight 2-in. balls, sixty-four 1-in. balls, and over 500 ½-in. balls. In practice, and in the same mill, 2-in., 1-in. and ½-in. balls of the same total weight as one 4-in. ball will strike 8, 64 and more than 500 blows respectively as will one 4-in. ball, and as a lower impact weight is ample

for the smaller particles, this smaller media becomes much more efficient and produces a much greater number of impact blows on the smaller part cles.

Fig. 2, reproduced from the same source but re-drawn to suit the conditions of the mill with coned lining, indicates a media classification for dealing with assorted rotary clinker having a maximum size of ⅜ in. to ½ in., and producing the finest commercial cement now made. The same classification should also be suitable for a cement of somewhat coarser residue ; in any case the mill's product would be consistent whether the cement was ground fine or rather coarse. In this diagram the net length of the mill is shown divided into ten equal sections, and, assuming the mill will require a total of 42 tons of media, the weight of the separate sizes would be as stated in the appropriate column of the following table.

ANALYSIS OF CHARGE MEDIA

Diameter (inches)	Per cent.	Net weight (tons)	Cumulative Per cent.
4 (approx.)	10	4·2	10
3 (approx.)	5	—	—
2½ and 2¼ mixed ..	5	4·2	20
2, 1¾ and 1½ mixed ..	10	4·2	30
1¼ and 1⅛ mixed ..	10	4·2	40
1 and ⅞ mixed ..	10	4·2	50
¾ and ⅝ mixed ..	10	4·2	60
½ (approx.)	10	4·2	70
7/16 and ⅜ mixed ..	15	6·3	85
5/16 and ¼ mixed ..	15	6·3	100
		42·0	

With regard to the small size of some of the media, it may be stated that the wear of this is at a very low rate ; as a matter of fact, it will be about only one-quarter that of the larger elements, perhaps owing to the less gritty nature of the material being ground.

The article referred to stated that the media might be charged into the mill indiscriminately, as it would sort itself out in due course ; but it appears best to charge the media into, say, two manholes—the largest at the feed end of the mill, and the smallest at or near the finishing end, and then leave the mill to do the rest. The lining plates at the finishing end of the mill would probably be of cast-iron or some brittle metal, and the trundling of the heavy media would probably cause fracture.

Experience with other types of lining indicates the necessity of the girthwise surface of the lining plates being uneven, notched, or surfaced in such a way that slipping of the media-grit mass is definitely prevented, and that this characteristic should be maintained until the plates are worn out ; the design of this should be carried out when the cone form is being dealt with.

Fig. 1 referred to a mill 8 ft. dia. by 24 ft. long, and as twelve rings of plates were shown it would appear that the plates were about 2 ft. long. The slope of

the cone surface is 1 in 3¾, and the mill is stated to have been satisfactory with this slope. A note in connection with the smaller mill stated that its slope was 1 in 12 and that classification was also satisfactory, but as the larger mill was in commercial use it appears best to base any work or design upon that. It will be noted that the slope of the last ring of plates is reversed, possibly with the object of encouraging the smallest media and the chips and broken pieces of media, etc., to keep away from the face of the diaphragm and so keep the face of the diaphragm and the slotted area clean.

Effect of the Size of Grinding Media on the Properties of Cement.

SOME experiments by Dr. F. O. Anderegg to determine the effect of the size of the grinding medium upon the properties of cement are described in "Rock Products" for May 1944. Three lots of clinker were ground using balls ⅞ in. and ⅜ in. in diameter. The mill had a diameter of 10 in. and a length of 12 in. It rotated at 40 r.p.m. and was charged with 100 lb. of balls and 10 lb. of clinker which had passed a mesh screen. The clinker had the following composition :

SiO_2	Fe_2O_3	Al_2O_3	CaO	MgO	SO_3	Insoluble Residue
21·20	3·28	6·06	63·0	2·24	1·69	0·12

The glass composition was estimated at about 5 per cent. The size distribution was determined in a butanol elutriator. It was soon observed that the smaller balls had difficulty in handling the very large grains, which often had a diameter of nearly ¼ in., so it seemed logical to compare the effect of grinding with ⅞-in. balls only with that of first grinding with the large balls followed by the smaller balls. On plotting the size distribution it was found that all samples were characterised by two apparent humps, the one at 10 microns and the other at from 25 to 30 microns. These humps were probably worse with the laboratory mill than would have been the case in a large tube mill with its greater number of impacts.

Workability and water retention were appreciably better after grinding with the smaller balls. An experiment was set up to determine the resistance to drying of the cement ground for 15 minutes with the large balls compared with that of the same clinker ground for 5 minutes with the large balls and for 10 minutes with the small balls. Mortar was mixed (1 : 3 with local river sand), and gauged to a flow of 105 per cent. with these two cements. The mortars were stored in a large bucket where some loss of water could occur by evaporation. Every 15 minutes a flow was run. At the end of an hour the mortar made with cement ground with large balls had a flow of only 45 per cent., while the other cement partly ground with small balls had a flow of 70 per cent., which was maintained for another half an hour. This seemed to indicate a better packing, with smaller capillaries between the grains to hinder water evaporation. The importance of this resistance to evaporation, especially in road work, is obvious.

Mortar strengths with standard sand at normal consistency were observed up to seven days. The strengths were appreciably higher when two sizes of balls were used (22 per cent. at 7 days) than with only the large balls. The strengths did not correlate with the water requirement, since this depends upon the surface area exposed and upon the voids between the cement grains. A larger surface requires more water to hydrate the cement to reduce strength. Poor size distribution means, on the other hand, interference with proper packing and more voids to fill, with decrease in workability, poorer water retention, and decreased strength. So the water requirement gives only ambiguous information under these conditions. The two lowest water requirements accompanied the two lowest strengths, while the cements requiring the greatest amount of water ranked sixth, fifth, and first in strengths. The humps in the grading curve undoubtedly contributed to the results observed, but the accuracy of the results obtained with the butanol elutriator in combination with the small laboratory mill were not sufficient to permit much accurate analysis.

These experiments need to be repeated on a large scale with a full size tube mill working on very uniform feed. Some data were obtained with $\frac{3}{8}$-in. balls in a large mill, but variations in amount and character of the clinker made it impossible to get results sufficiently good to draw discriminating conclusions. However, it was evident that finishing with the smaller balls produced a cement appreciably superior in working qualities and strength for a given amount of energy expended.

Some results obtained in grinding with air separation were found to be of interest in this connection. In one case clinker with a surface area of about 640 sq. cm. per gramme was ground in a 7 ft. by 22-ft. tube mill loaded to 45 per cent. of its volume with $\frac{7}{8}$-in. balls. After separation its area was about 1,700 sq. cm. per gramme. On plotting a size distribution curve, a considerable hump at about 25 microns was apparent. On regrinding a similar cement in the same type of mill and with similar media, a surface area of about 2,700 sq. cm. per gramme was obtained and the cement had a small hump at 20 microns. Whereas a yield of 90 barrels per hour was obtained on straight grinding, with a surface increase of about 1,000 sq. cm. per gramme, the regrinding gave only a little more surface increase (about 1,100), but the yield was only 47 barrels per hour.

The efficiency of grinding falls off greatly in making high-early-strength cements. It is suggested that decreasing the size of the balls, with a corresponding increase in surface and number of impacts, will tend to give greater efficiency and probably help to give a smoother grading curve.

Heat Economiser for the Wet and Dry Process.

IN the article on this subject in our number for November, 1944, it was stated on page 104 that the guaranteed coal consumption was 1,630 kilo calories per kilo of clinker. This should read 1,673 kilo calories per kilo of clinker.

Heat Efficiency of a Cement Kiln and Waste-Heat Boiler.

BY permission of the South Australian Portland Cement Co., Ltd., Mr. H. H. Wight, B.Sc., Engineer at the Brighton, South Australia, works of the Company, has published in the "Commonwealth Engineer" the following data relating to the heat efficiency of a rotary kiln with a waste-heat boiler. The kiln (by Messrs. F. L. Smidth and Co., Ltd.) is 157 ft. long by 8 ft. 6 in. internal diameter reduced by a brick lining 7 in. thick. The boiler has 6,843 sq. ft. of heating surface and the economiser has horizontal tubes. It is pointed out that a number of quantities cannot be measured by meter at present, and an accurate evaluation of boiler efficiency is not possible, but that by completing a heat balance on the kiln a fair accuracy has been secured.

The following figures represent conditions applying when the kiln is running smoothly at a daily output of 133 tons of clinker during 23 hours of operation (5·8 tons per hour). The whole plant is then operating on "works power."

Load on generator, 620 kW. = 830 h.p. = 83 per cent. of the capacity of the set.
Boiler pressure, 170 lb. per square inch.
Steam temperature, 405 deg. F. = 30 deg. superheat. $H = 1{,}217$.
Engine pressure, 150 lb. per square inch.
Steam temperature, 390 deg. F. = 24 deg. superheat. $H = 1{,}211$.
Gas temperature at kiln outlet, 1,500 deg. F. = 1,960 *FA* = 815 deg. C.
Gas temperature at boiler inlet, 1,450 deg. F. = 1,910 *FA*.
Gas temperature at economiser inlet, 470 deg. F. = 930 *FA*.
Gas temperature at economiser outlet, 350 deg. F. = 810 *FA* = 177 deg. C.
Boiler feed rate, 20,000 lb. per hour.
Engine feed rate, 17,000 lb. per hour.
Cooling pond, 108 deg. F.
Jet condenser exit circulating water, 138 deg. F.
Partial air pressure, 0·6 in.; barometric, 29·7 in.; vacuum, 23·5 in.
Thermal efficiency of engine, condenser unit

$$= \frac{620 \times 3{,}415}{17{,}000\ (1{,}211 - 76)} = 11 \text{ per cent.}$$

A modern turbine installation of the same power would have an efficiency of about 17 per cent.

The efficiency of the entire plant as a power station is

$$\frac{620 \times 3{,}415}{4{,}280 \times 0{\cdot}977 \times 12{,}500} = 4{\cdot}0 \text{ per cent.}$$

The efficiency of the boiler-economiser unit has been greatly improved by (1) Adding sets of steam-lance shutters as required until no accumulation of flue dust can occur in a position from which it cannot be blown by a lance. (2) Varying the types of lances to suit (1). (3) Eliminating unnecessary hangers, etc., near the superheater to reduce "accumulation zones" to a minimum. (4)

Connecting lance steam lines on to the dry steam main. (5) Dispensing with soot-blowers on account of low superheat and caking of flue dust. (6) Installing gauges across each pass of the unit to enable attendants to watch the condition of each section of the tubes.

GAUGE READINGS

Primary air fan	+ 18·00	in.	W.G.
Kiln outlet	− 0·08	,,	,,
Boiler inlet	− 0·28	,,	,,
No. 1 pass drop	− 0·65	,,	,,
No. 2 pass drop	− 0·45	,,	,,
No. 3 pass drop	− 0·45	,,	,,
No. 4 pass drop	− 0·60	,,	,,
Economiser drop	− 0·70	,,	,,
Economiser outlet	− 3·13	,,	,,
Dust collector drop	− 0·34	,,	,,

The raw meal feed controller was set to give 11,000 revolutions per shift of the two feed screws. The secondary air fan controller was on position 10. The primary air fan suction was full open. The coal feed controller was set to give 23,000 revolutions per shift of the two feed screws. The kiln speed controller was set to turn the kiln at the rate of 410 revolutions per shift (67 seconds per revolution). The operating day was 23 hours, and each shift was 7·40 hours.

Boiler-economiser efficiency depends on an accurate measurement of inlet heat to boiler, and so a heat balance on the kiln is set out to obtain the required check.

GAS-FLOW COMPUTATIONS. Kiln exit, 1,500 deg. F.

Fuel analyses (as fired):

Approximate Analysis.

VHC	FC	ASH	H_2O	S
30·4	53·6	14·7	2·3	1·8

Ultimate Analysis (dry basis).

C.V.	S	H	C	O	N
12,500	1·8	4·9	69·2	6·7	1·3

Raw Meal: $CaCO_3$, 76·4 per cent.; $MgCO_3$, 3·8 per cent.; $Al_2Si_2O_7$ (hyd.), 8·7 per cent.; H_2O, 0·2 per cent.

C_aCO_3 is 44 per cent. CO_2, M_gCO_3 is 52.4 per cent. CO_2. In 100 lb. of raw meal, there are (33·6 + 2) = 35·6 lb. CO_2.

Flue Dust: H_2O, 0·5 per cent.; CO_2, 5·5 per cent.; C, 0·5 per cent.

Rates of Flow:

133 tons of clinker from 205 tons feed or raw meal = 20,000 lb. per hour.

16·5 tons of flue dust, 1,610 lb. per hour.

∴ Kiln feed, 221·5 tons per day = 21,600 lb. per hour.

Coal, 33 per cent. of clinker = 1·91 tons per hour = 4,280 lb. per hour.

Ash, 4,280 × 14·7 = 6·5 tons per day = 630 lb. per hour. (Assume that all the ash is entrained in flue gases.)

Flue-dust returned from dust collector, 10 tons per day = 975 lb. per hour.

Dust collector and other flues, 5/3 and total flue dust = 16·5 tons per day if stack loss is 3·5 tons per day.

Therefore, kiln feed loss to flue gases is (approx.) 13·5 tons per day = 6·1 per cent. of feed.

Ash, 630/21,600 = 2·92 per cent. of feed.

Therefore, assuming equal proportions of ash and raw meal are entrained (and also deposited), and that the stack loss is 3·5 tons per day.

Therefore, 20 tons per day lost to flue gases = 6·5 ash and 13·5 kiln feed, that is, 32·5 per cent. ash, 67·5 per cent. kiln feed. Thus, returned flue dust = 32.5 per cent. ash and 67·5 per cent. partially de-carbonated kiln feed.

Therefore, 133 tons of clinker come from (221·5-13·5) = 208 kiln feed clinker = 63·9 per cent.

Thus the kiln feed of 221·5 is 205/221·5 = 92·6 per cent. raw meal and 7·4 per cent. flue dust. Therefore of the 13·5 tons of feed entrained by flue gases 12·5 are from raw meal, and 1 ton from flue-dust (205 + 6·5) = 211·5 in = (208 + 3·5) = 211·5 out.

Gases from Kiln Feed : From raw meal : Co_2 = 20,000 × 0·356 = 7,120 lb. per hour. H_2O = 20,000 × 0·002 = 40 lb. per hour. Less losses in stack dust :

CO_2, (3·5 × 2,240/23) × 0·055 = 18·7 lb. per hour. H_2O = (3.5 × 2,240/23) × 0·055 = 1·7 lb. per hour. Therefore CO_2 into flue gases = 7,100 lb. per hour. H_2O into flue gases = 38·3 lb. per hour.

Assuming primary air at 80 deg. F. and 50 per cent. humidity, from Goodenough's tables, dry air = 13.60 cu. ft. per lb. and saturated air = 14·09 cu. ft. per lb. Therefore with 50 per cent. humidity = 13·845 cu. ft. per lb. and moisture of dry air per lb. = 0·02226/2 = 0·01113 lb.

Therefore at 80 deg. in 1 lb. of humid air there are 13·845/1·0111 = 13·69 cu. ft. and in 1 lb. of humid air there are 0·01113/1·0111 = 0·0110 lb. of vapour.

Therefore by weight the air contains 1·10 per cent. water vapour, by volume, the air contains 0·245/13·845 = 1·77 per cent. water vapour.

Composition of Air :

	Per cent. by volume	Per cent. by weight	P.p. cu. ft. N.T.P.
N_2	77·40	75·70	12·77
O_2	20·60	22·85	11·21
H_2O	01·77	01·10	—
CO_2	00·23	00·35	8·15
SO_2			5·60
CO			12·81
Dry air (N.T.P.)			12·39
Primary air 80 deg. F., 50 per cent. humidity)			13·69

Gases from Combustion of Fuel :

	S	H	C	O	N
Per cent. of weights	1·8	4·9	69·2	6·7	1·3
Lb. per hour	75·2	205·0	2,890·0	280·0	54·4

Total, 83·9 per cent. ; dry coal, 4,180 lb. per hour.

These elements burn as follows :

	S	H	C
Lb. per hour	SO_2-150	H_2O-1,845	CO_2-10,600

requiring the following amounts of O_2 :

Lb. per hour	75	1,640	7,710

that is a total of 9,425, less 280 = 9,145 lb. per hour, and this is obtained from 40,000 lb. of air per hour.

Lb. per hour :

N_2	CO_2	H_2O	SO_2	O_2	
54	10,600	1,845	150	280	(from coal)
30,280	140	440	—	9,145	(from air)
—	7,100	38	—	—	(from raw meal)
30,330	17,840	2,323	150		utilised totals.

The water vapour will all be true steam, and the lb. per cu. ft. have been taken from an H-ϕ diagram.

Temperature,	1,500 deg. F.
also,	350 deg. F.
Lb. per cu. ft.	73
	− 31

1,500 deg. F. temperature factor = 1,960/492 = 3·99.

Converting to cubic feet per hour :

N_2	CO_2	H_2O	SO_2	O_2
1,544,000	580,000	170,000	3,350	—

∴ CO_2 per cent. from Orsat = 580/2,127 = 27·3 (theoretical maximum).

The SO_2 is only included to make these representative calculations more complete and would be absorbed with the CO_2.

If the coal had been fired in a boiler, the Orsat reading for maximum CO_2 per cent. becomes 350/1,897 = 18·5.

VARIATION DUE TO INCOMPLETE COMBUSTION :

Assume CO per cent. from Orsat = 0·6. Therefore cu. ft. per hour approx. = 12,800 at 1,500 deg. F. = 250 lb. per hour, which comes from 107 lb. per hour C. Therefore C burnt to CO_2 = 2,890 − 107 = 2,783, which gives 10,200 CO_2, requiring 7,420 of O_2.

The 250 lb. per hour CO require 143 of O_2. Therefore, total oxygen required is 75 + 1,640 + 7,560 = 9,275, less 280 = 8,995 lb. per hour, and this is now obtained from 39,350 lb. of air per hour.

Lb. per hour :

N_2	CO_2	H_2O	SO_2	O_2	CO	From
54	10,200	1,845	150	280	250	coal
29,800	138	433	—	8,995	—	air
—	7,100	38	—	—	—	raw meal
29,850	17,440	2,316	150	utilised	250	

Cu. ft. per hour :

1,520,000	567,000	169,000	3,350	—	12,800

Orsat per cent. :	N_2	CO_2	CO	SO_2
	72·3	26·9	0·6	0·14

VARIATION DUE TO EXCESS AIR :

Assume O_2 per cent. from Orsat = 0·8 ; cu. ft. per hour = 16,850 (approx.) which = 377 lb. per hour.

Lb. per hour becomes :

N_2	CO_2	H_2O	SO_2	O_2	CO	From
54	10,200	1,845	150	280	250	coal
29 800	138	433	—	8,995	—	air
1,250	7	18	—	377	—	excess air
—	7,100	33	—	—	—	raw meal
31,100	17,440	2,334	150	377	250	

Cu. ft. per hour :

1,585,000	567,000	170,000	3,350	16,850	12,800

Orsat per cent. :	N_2	CO_2	O_2	CO	SO_2
	72·5	25·9	0·77	0·60	0·14

More accurately, the oxygen would be 0·8 per cent. of the latter figures, which brings it to 17,500 cu. ft. per hour. The corrected totals are then :

1,587,000	567,000	170,000	3,350	17,500	12,800	Totals = 2,358,000

Lb. per hour :

31,150	17,440	2,335	150	391	250	= 51,720

= 3·98 lb. per lb. of clinker, and 12·10 lb. of fuel (1·67 from raw meal).

HEAT CONTENT OF FLUE GASES :

The following formulæ for the specific heat of gases at temperatures t deg. F. have been chosen after comparing the formulæ given by several authorities :

$$N_2 \text{ or } CO—0{\cdot}2421 + (0{\cdot}000\ 0119\ t - 32)$$
$$CO_2—0{\cdot}192 + (0{\cdot}000\ 0054\ t - 32)$$
$$SO_2—0{\cdot}117 + (0{\cdot}000\ 0617\ t - 32)$$
$$O_2—0{\cdot}2119 + (0{\cdot}000\ 0104\ t - 32)$$
$$\text{Water vapour } —0{\cdot}433 + (0{\cdot}000\ 0967\ t - 30)$$

Therefore the mean specific heats to 1,500 deg. F. are : N_2 or CO, 0·2508 ; CO_2, 0·196 ; SO_2, 0·162 ; O_2, 0·2195. Therefore the heat contents above 32 deg. F. are :

		B.Th.U. per hour
N_2 — 31,150 × 0·2508 × 1468	=	1,147 × 10^4
CO_2 — 17,440 × 0·196 × 1468	=	501
H_2O — 2,335 (1,151 + 634)	=	417
SO_2 — 150 × 0·162 × 1,468	=	3·6
O_2 — 391 × 0·2195 × 1,468	=	12·7
CO — 240 × 0·2508 × 1,468	=	9·2
Total		2,091 × 10^4

Therefore the mean specific heat of flue gas = (2,091 × 10^4)/51,720 × 1,468) = 0·276. Similarly the mean specific heat at 35 deg. F. = 0·388, but this will be affected by additional air leaks and steam from lances.

HEAT BALANCE OF KILN

HEAT GAINS ABOVE 32 DEG. F. :

	B.Th.U. per hour$_1$
Calorific value of fuel, 4,280 × 0·977 × 12,500 =	52·2 × 10^6
Sensible heat in moist air drawn into kiln, 9,372 × 100/22·85 × 0·238 (80 — 32) =	0·47
Sensible heat in dry kiln feed, (20,000 × 0·998 + 1610 × 0·005) 0·21 (142 — 32) =	0·46
Heat liberated by formation of clinker, 13,000 × 180$_1$ =	2·34
Heat liberated by formation of clinker, 13,000 × 180· =	2·34
Friction heat generated in kiln by rolling of feed, 15 × 0·746 × 3,416 =	0·04
Total	55·5 × 10^6

HEAT LOSSES ABOVE 32 DEG. F. :

Heat in exit flue gases .. .. =	20·91 × 10^6
Heat to decompose $CaCO_3$, (20,000 × 0·764 × 710) + (1,610 × 0·055 × 100/60 × 76·4/80·2 × 710) =	10·95
Heat to decompose $MgCO_3$, (20,000 × 0·38 × 610) + (1610 × 0·055 × 100/60 × 3·8/80·2 × 610) =	0·47
Heat lost in discharged clinker, 13,000 × 0·28 (950 — 32) =	3·32
Heat lost by production of CO, 10,200 × 250 =	2·55
Loss from combustible in flue dust, 1,610 × 20·0/16·5 × 0·005 × 12,500 (approx.) =	0·12
Heat lost in discharged flue dust, 1,610 × 0·25 (1,500 — 142) =	0·55
Heat lost in stack dust loss 341 × 0·25 (1,500 — 32) =	0·13
Heat lost in evaporating moisture in coal, 4,280 × 0·023 × 1,785 =	0·18

Heat loss by unburnt H_2 — assumed 50 per cent. of CO loss (not included in gas computations) = 1·28

Heat to dehydrate kaolin, 20,000 × 0·087 × 75 = 0·13

Heat loss in evaporating moisture in raw meal, 20,000 × 0·002 (1,770 − 142 + 32) = 0·07

Latent heat of steam from combustion of H_2 credited to calorific value, 4,280 × 0·977 × 0·049 × 18/2×971 = 1·79

(Sub-total) 42·4

HEAT LOSS BY SHELL RADIATION :

$E = eo\text{-}\ (\theta^4 - es\theta s^4)$. Research has shown *es* to be 1·00 for low temperatures and independent of colour.

$o\text{-} = 5{\cdot}7 \times 10^{-12}$ watts per cm.^{2} (C.G.S.)

$= 17{\cdot}2 \times 10^{-10}$ B.Th.U. per sq. ft. per hour (F.P.S.)

The kiln has been divided into five sections :

Length (ft.)	Area (sq. ft.)	θ	θs	e	E
57	1,539	640	545	0·90	0·10 × 10^6
40	1,080	700	545	0·85	0·24
37	999	800	550	0·80	0·44
13	351	1,200	560	0·70	0·84
11 coolers	830	1,200	560	0·70	1·98
Kiln hood	24	1,000	555	0·75	0·03
					Total 3·72 × 10^6 (6·7 per cent. of total)

HEAT LOSSES BY CONDUCTION TO AIR.—The author has found that this has been expressed by investigators in a wide range of formulæ. (1) Griffiths and Davis have used

$Hc = C(t - t) - {}^5/{}^4$ = B.Th.U per sq. ft. per hour plate air

where $C = 0{\cdot}32$ for normal range of conditions, and temperatures are in degrees F.

(2) J. E. Bell has arranged a formula of Osborne Reynolds, $Hc = R(t - t)$ plate air, where R = about 4 for the above conditions.

The author again used the above five sections of kiln and set out figures on actual flow as follows :

Length	Air stream width	Air speed ft. per sec.	Cu. ft. per hour	Air temp. deg. F.	Rise temp. deg. F.	Lb. per hour	B.Th.U. per hour
57	3·3	1·0	677,000	135	50	44,900	0·54 × 10^6
40	3·3	1·4	664,000	155	70	42,600	0·72
37	3·3	2·4	1,055,000	210	120	62,100	1·81
13	3·3	3·0	464,000	245	150	26,000	0·95
15	6·4	4·0	1,013,000	300	200	49,400	2·42
						Total	6·44 × 10^6

Of the two formulæ quoted above (1) gives a definitely lower result whereas (2) totals close to this figure. Accordingly,

Losses by conduction to air, $5{\cdot}0 \times 10^6$. Therefore

$$\text{total heat losses} = \begin{array}{r} 42{\cdot}5 \\ 3{\cdot}7 \\ 5{\cdot}0 \\ \hline 51{\cdot}2 \end{array} \times 10^6 \text{ B.Th.U.}$$

That is, the heat balance closes to $51{\cdot}2/55{\cdot}5 = 92{\cdot}3$ per cent., which is satisfactory since no attempt has yet been made to close this balance.

The missing $4{\cdot}3 \times 10^6$ heat units are now allocated:

Heat in exit flue gases,	$20{\cdot}91 + 1{\cdot}90 = 22{\cdot}8 \times 10^6$
Heat in discharged clinker,	$3{\cdot}32 + 0{\cdot}50 = 3{\cdot}8$
Heat in discharged flue dust	$0{\cdot}55 + 0{\cdot}10 = 0{\cdot}65$
Heat in stack dust loss,	$0{\cdot}13 + 0{\cdot}05 = 0{\cdot}18$
Radiation loss,	$3{\cdot}7 + 0{\cdot}75 = 4{\cdot}45$
Conduction loss,	$5{\cdot}0 + 1{\cdot}00 = 6{\cdot}0$
	$4{\cdot}3$

BOILER-ECONOMISER EFFICIENCY

$$= 18{,}500\ (1{,}217 - 140 + 32)/22{\cdot}8 \times 10^6 = 89{\cdot}9 \text{ per cent.}$$

(The author is aware that this percentage should probably be in the region of 83 per cent., but no certain factor could be adjusted on account of particularly weak assumption, and the high figure has been allowed to remain.)

ECONOMISER EXIT GAS CALCULATIONS—350 DEG. F.

	N_2	CO_2	H_2O	O_2	CO	(neglected SO_2)	
Cu. ft. per hour	1,587,000	567,000	170,000	17,500	12,800	1,500° F.	Total
	∴ 655,000	234,000	72,200	7,230	5,290	350° F.	974,000
Plus steam from two hand lances = 1,300 lb. per hour = 40,300 cu. ft. per hour.							
cu. ft. per hour	655,000	234,000	112,500	7,230	5,290	Total	1,014,000

Let L cu. ft. per hour be the total air leakages,

then

$$\begin{aligned} N_2 &= 655{,}000 + 0{\cdot}774L \\ CO_2 &= 234{,}000 + 0{\cdot}0023L \\ H_2O &= 112{,}500 + 0{\cdot}0177L \quad \text{Orsat total: } 902{,}000 + 0{\cdot}982L \\ O_2 &= 7{,}230 + 0{\cdot}206L \\ CO &= 5{,}290 \end{aligned}$$

$\therefore CO_2$ per cent. $= (234{,}000 + 0{\cdot}0023L/(902{,}000 + 0{\cdot}982L)$.

Actual tests by Orsat gave $15{\cdot}6$ per cent. CO_2; therefore L = air leakage per hour = 619,000 cu. ft. = 61 per cent.

Approximately $L = (25{\cdot}9 - 15{\cdot}6/15{\cdot}6 = 66$ per cent. Therefore, total exit gas = 1,633,000 cu. ft. per hour = 27,200 cu. ft. per minute.

The O_2 should be $(7{,}230 + 0{\cdot}206N)/(902{,}000 + 0{\cdot}982N) = 8.93$ per cent., and this checked very closely with test figures. The Orsat now reads :

N_2	CO_2	O_2	CO
75·1	15·6	8·9	0·35

POSSIBILITY OF CONDENSATION IN EXIT FLUES :

Lb. per hour of dry gas = 49,300
29,800

79,100

Lb. per hour of steam = 3,630 (from kiln and lances)
340 (air leaks)

3,970 = 5·02 per cent.

Therefore the gases will not become saturated until the temperature falls to 105 deg. F. Absorption of moisture by CaO in dust collector = 975 = 75 per cent. × 18/56 = 235 = 5·9 per cent. of total.

Accelerated Test for Cement Manufacture.

ACCORDING to " Zement," a method of accelerating the hardening of test mortars to allow compressive and bending (tensile) strengths of clinker and cement to be determined after a few hours has been developed by F. W. Meier-Grolman and W. Biek. An abstract of an article in " Zement " is published in " Building Science Abstracts," where it is stated that the standard mortar in the mould is placed in an electrically-heated drying closet at 135 deg. C. and stored at that temperature for 2½ hours. The specimen is then removed from the mould and tested immediately. Satisfactory results are obtained. From the results of the accelerated strength test, the boiling test, and the determination of fineness, an accurate conclusion respecting the quality of a cement can be reached.

Cement Production in the United States.

THE downward trend of the production of Portland cement in the United States, which became very pronounced as the war-time constructional requirements came to an end, showed an upward tendency during the earlier months of last year. Figures issued by the United States Bureau of Mines show that the ratio of production to capacity was 29 per cent. in February last year, 29 per cent. in March, 32 per cent. in April, and 35 per cent. in May. In May, 1943, the volume of production was 59 per cent. of total capacity.